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MILLING METHODS AND COSTS AT THE HARMONY MINES, BAKER, IDAHO

BY

R. DUNCAN GARDNER

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MILLING METHODS AND COSTS AT THE HARMONY MINES, BAKER, IDAHO[1]

By R. Duncan Gardner[2]

INTRODUCTION

This paper describing the milling practice at the concentrator of the Harmony Mines Co. is one of a series being prepared by the Bureau of Mines on milling methods and costs in the United States.

For the period February 15, 1929, to June 1, 1929, the concentrator treated 200 tons of copper ore per day containing 3.84 per cent copper and recovered 99.2 per cent of the total copper in concentrates averaging 28.61 per cent copper.

Milling practice at the Harmony Mines is in most respects similar to the usual practice at the average flotation concentrator producing bulk flotation concentrates. The exceptionally high recovery is due to a number of minor conditions. These minor details will be stressed throughout this paper.

LOCATION

The concentrator is 8 miles up Withington Canyon from Baker, Idaho. Baker is situated on the Gilmore & Pittsburg Railroad.

The mill is built on a hill side having a 25° slope, and the machinery is so arranged that elevation of products is not necessary, except in one instance where additional storage capacity is so obtained.

The concentrator employs the flotation process and is designed to operate with the least possible power and labor. Water is obtained from a spring some distance above the mill and reaches the storage tank through a 500-foot pipe line. Electric power is transmitted 3 miles from a Diesel-engine power plant at Baker. The power is generated at 2,300 volts, stepped up to 23,000 for transmission and stepped down to 2,300 at the mine. All motors in the concentrator and at the mine operate at this voltage.

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2 Consulting engineer, U. S. Bureau of Mines.

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ORE TREATED

The ore comes from the Harmony mine, which is located up Withington Canyon about half a mile beyond the concentrator. A shrinkage system of mining is used, and the ore is sent to the concentrator over an aerial tramway.

Copper is the only valuable metal in the ore. Small amounts of gold and silver are also present but in such small quantities that they are insignificant even in the concentrates. The copper content of the ore reaching the mill varies from $2\frac{1}{2}$ to 5 per cent. Some testing, however, was done with ore running as low as $\frac{3}{4}$ per cent copper.

The most important copper mineral in the ore is chalcopyrite, although chalcocite and bornite are found in the upper levels of the mine. During the period covered by this report chalcopyrite was the only mineral in the ore treated.

The ore occurs in stringers of various widths in sheer zones of an impure quartzite. In all the stringers the chalcopyrite is intimately mixed with pyrite. The relative amounts of the two minerals vary to a considerable degree, but even in hand-picked specimens of chalcopyrite pyrite is always present. The stringers or veinlets are separated or inclosed by extremely hard, shaly quartzite. The rock not only has the hardness of quartz but is exceptionally tough as well, the small percentage of shale having the effect of cementing the particles into a tough mass. Because of these characteristics the ore is very hard to crush and causes excessive wear on all metal parts of machines coming in contact with it. Some serious damage has also been done to crushers due to the difficulty of breaking down the larger chunks of ore.

The shape of the ore body is somewhat irregular, without definite walls, consequently barren material is broken with the ore. Efforts to hand-sort the ore and remove this barren portion proved more expensive than the cost of milling the mine ore. A very small percentage of the waste is removed in this way.

As it is broken in the stopes the ore is in rather large pieces. These are broken further in the bulldozing chambers to pass 18-inch grizzly openings. The ore passes over another grizzly with 12-inch openings at the head house of the aerial tramway, where sledge hammers are used in breaking up the large pieces.

The moisture content of the ore is less than 1 per cent, although water is present in all the lower stopes. The lack of moisture is due to the solid condition of the veins and the compactness of the ore.

The ore is distinctly sulphide in character and shows only a slight tendency toward oxidation after being broken. It is noticeable that where the broken ore has remained in stopes for several months it takes on a rusty tinge, due to oxidation of the surface of the mineral crystals. This evidence of oxidation is more noticeable where water has dripped through the mass of broken ore.

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HISTORY OF CONCENTRATOR OPERATIONS

The concentrator, which was built in 1917, was originally designed to treat the ore by gravity concentration, and for several years this method was followed. The flow sheet was somewhat complex, and numerous changes were made to increase the efficiency of the plant. High recovery was never achieved while the gravity method of concentration was used. It was also found quite impossible to effect separation of the pyrite and the copper-bearing minerals by this method. The concentrates were therefore not desirable for the smelters, since they carried too much sulphur and iron in comparison to the copper content.

Flotation was added in 1922 as a supplementary unit to treat all the slimes and some table middlings. This arrangement slightly increased the recovery, since it saved a portion of the copper which was being lost in the slimes. The grade of concentrates, however, was not changed materially, and the recovery was still unsatisfactory.

Table concentration was discontinued in 1928, and some test runs were made using straight flotation. The milling machinery was not rearranged for these tests, and it was found necessary to elevate the pulp several times before concentration was accomplished. The results of these tests were not wholly satisfactory, but the poor results were due more to the awkward arrangement of the concentrating machinery than to any fault in the concentrating method used.

In December, 1928, a complete rearrangement of concentrating machinery was decided upon. The most important of the changes was the moving of the flotation cells as close to the ball mill as possible, thus making the concentrator more compact by eliminating elevators and eliminating three-fourths of the launders formerly used. The flotation cells had previously been situated on the opposite side of the table floor from the ball mill, a distance of nearly 100 feet.

The use of straight flotation and the rearrangement of machinery increased the mill recovery more than 20 per cent while the grade of concentrates was raised from an average of 20 per cent to 28 per cent copper. The labor cost of concentrating was cut in half, and power consumption was reduced by 30 per cent.

Other advantages that proved of great importance were greater flexibility of operation and closer adjustment between the various reagents that were used in the process. Tests and experiments made in the laboratory on a smaller scale were helpful in perfecting the process.

General conclusions as to the reasons for high recovery and clean concentration of the ore under consideration are that they depend on the exercise of extreme care in carrying out the details of the process rather than on any innovation or wide departure from customary practice. For instance, the use of too much or too little reagents, a wide variance in pulp density, and overloading the ball mill or flotation cells cause greater irregularities and losses than the use of a slightly different reagent or the fact that the pulp is acid or alkaline.

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A series of test runs made after the machines had been installed in the new positions, together with some carefully made observations, brought out some important facts. Some of these are:

1. Best recoveries are obtained when the ore is moved quickly from the stopes to the concentrator.

2. The feed to the ball mill must be as fine as practicable; and, most important, it must be uniform in size for best results.

3. The finer the ore is ground in the ball mill the higher the recovery in the flotation cells.

4. To make a clean separation of chalcopyrite and pyrite and at the same time obtain a high recovery, both rougher and cleaner flotation cells must be used. In practice, the tailings end of the rougher cell is kept free from copper minerals and only clean concentrates are taken from the cleaner cell.

5. Cleaner-cell tailings are not sent back to the rougher cell but pumped to the head of the ball mill and ground to a finer size.

6. Excessive amounts of reagents lower the grade of concentrates.

7. Agitating the pulp between the ball mill and the flotation cells has an undesirable effect upon flotation-cell operation.

8. The direction of movement of the pulp in the frothing chamber must be downward for best results.

9. A long flotation cell allows cleaner separation than a short one.

10. The density of the pulp is of the utmost importance, and a slight change in pulp density causes a great difference in flotation cell operation.

11. Concentrates from a drier or a filter should not be returned to the flotation cells.

12. All cleanings from cells, classifiers, filters, ball mill and launders should be sent to the head of the ball mill in as nearly a uniform stream as possible.

Flow sheet

Figure 1 gives the flow sheet of ore treatment. Figures 2 and 3 give a side view and plan of the concentrator, respectively, showing the position of the various machines.

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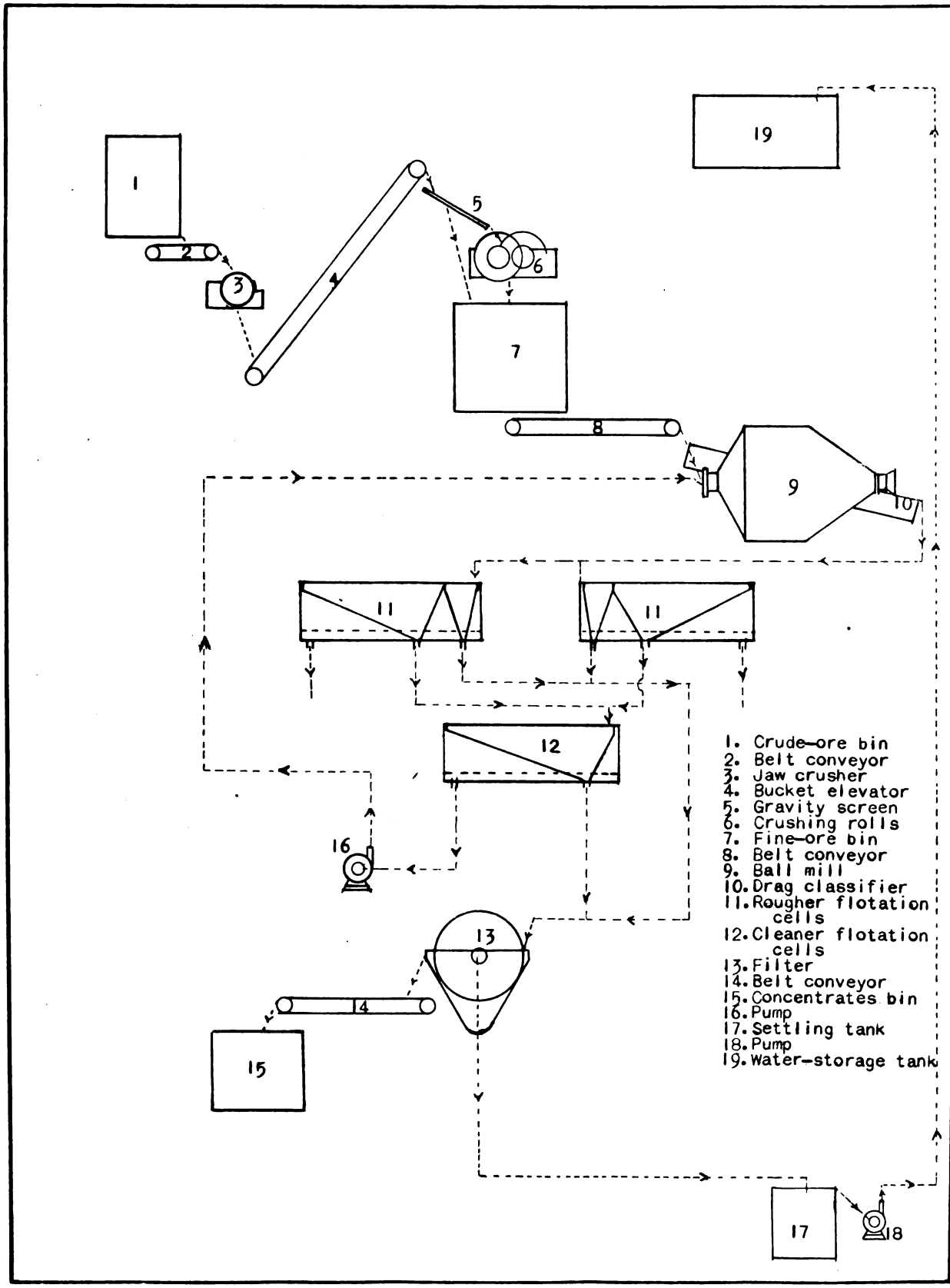

Figure 1.- Flow sheet of 200-ton concentrator

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The ore comes from the mine over an aerial tramway and is stored in a 200-ton-capacity crude-ore bin. A feeder delivers the ore to a belt conveyor, equipped with a magnetic pulley, which in turn drops the ore into the mouth of a Universal jaw crusher set to crush to 1½-inch size. A bucket elevator raises the crushed ore product to a gravity screen having ½-inch holes. The oversize of the screen is crushed by a pair of rolls set at ½ inch; the product from the rolls joins the undersize of the screen and is stored in a fine-ore bin of 400 tons capacity.

The ore from this bin passes through an adjustable steel hopper to an automatic belt feeder, which delivers the ore to a Hardinge ball mill in closed circuit with a Dorr drag classifier. The overflow from the classifier goes directly to the rougher flotation cells, and the sands from the classifier return to the ball mill. The flotation collecting reagent is added at the head of the ball mill, and the frothing reagent is added at the head of the rougher flotation cells.

The tailing of the rougher flotation machine flows to the tailings pond, and the rougher flotation concentrates are treated in the flotation cleaner cells, producing a middlings product that is pumped to the head of the ball mill and finished concentrates that are filtered and delivered by a belt conveyor to the concentrates storage bin. A wagon haul of eight miles delivers the concentrates to the railroad at Baker, and from this point the concentrates are shipped to the smelter.

Tramway

The aerial tramway which carries the ore from the mine tunnel level to the concentrator is 1,800 feet long and consists of a 1¼ inch traction cable and a 7/8-inch draw cable supported by three towers, the longest span being 1,200 feet. The tramway is equipped with 14 buckets, each of 8 cubic feet capacity. A 10-horsepower motor at the upper terminal is used to start the tram and acts also as a break during operation. A certain amount of power is thus forced back into the circuit, but no attempt has been made to ascertain the exact amount. The tram, when running at capacity, has a speed of 600 feet per minute, and the average amount of ore sent over it in eight hours exceeds 200 tons, although the capacity could be doubled by adding more buckets. Automatic trips operate the clutches which hold the buckets to the draw cable; loading and dumping of the buckets is done by hand.

Feeder

The ore bin at the lower end of the tramway has a capacity of 200 tons and is built with a steeply sloping bottom so that practically all of the ore will run from the chute without being barred down. The chute has an opening 26 inches wide and 36 inches high, and the flow of ore through this chute is regulated by an automatic feeder. The feeder is cylindrical and composed of six heavy vanes which fit into slots in two heavy end disks. It is turned by a 2-inch shaft, and its

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speed is regulated by varying the size of the driving pulley. The middle of the feeder is set 2 inches above the bottom of the chute, and due to its position ore cannot pass over it when it is stationary. The amount of ore passing over it is therefore dependent upon the speed of the feeder and the heighth of the gate opening in the chute. The chute bottom is covered with sheet steel to keep the ore from hanging up ahead of the feeder. The feeder drops the ore directly on a conveyor belt 24 inches wide and 6 feet between pulley centers operating at a speed of 60 feet per minute.

The conveyor belt carries the ore over a magnetic pulley which lifts out all tramp iron. This pulley is the common type with its magnets operating under a direct current at 110 volts, the current being supplied by a generator driven from the main line shaft.

Primary crushing

As previously stated, the ore drops from the conveyor belt into the hopper of a 16 by 18 inch jaw crusher set to crush to 1½-inch size. The crusher is driven by a belt from a line shaft above the floor. This drive has proved the most desirable, since it saves the motor from excessive shocks and also has the added advantage over geared drives that the belt will slip when the crusher is overloaded or receives material too resistant to be crushed.

The crusher is run at a capacity of approximately 200 tons in eight hours, but in cases of emergency double this amount has been put through the machine. The amount crushed is regulated entirely by the automatic feeder under the ore-bit chute, which keeps a uniform stream of ore flowing into the crusher and under normal conditions the crusher is seldom overloaded. Repairs to the unit were practically negligible during the period under consideration.

Intermediate crushing

Formerly a crusher of the same type as the primary breaker was used to crush the product of the primary crusher down to ½-inch size. The capacity of this crusher, however, when used to reduce the ore to the size stated was never greater than half that of the primary breaker, so that difficulty was experienced with this unit. A set of rolls 30 by 18 inch, having a capacity equal to that of the primary breaker, was installed to replace the secondary breaker.

The crushed product of the primary breaker passes over a 2½ by 5 foot gravity-type screen having ½-inch holes, the undersize of the screen amounting to approximately 35 per cent of the screen feed going to the fine-ore bin. The oversize, amounting to approximately 130 tons per day, goes to the rolls set to crush to ½-inch size, the crushed product of the rolls going to the fine-ore bin. The rolls have been found to give a much more uniform product than the secondary crusher formerly used.

All machines in the crushing department, both primary and secondary, are driven by a single 30-horsepower motor.

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Grinding

The fine-ore bin, of 400 tons capacity, has a specially constructed steel hopper at the bottom with adjustable gate which regulates the flow of ore to a 24-inch belt conveyor. The speed of this belt is varied by a cam and ratchet arrangement which pulls the belt ahead a variable fraction of the circumference of the drive pulley at each revolution of the main drive shaft. The amount of ore fed to the ball mill can thus be varied within the wide limits of from 2 to 80 tons per hour.

From the conveyor belt the ore drops into the ball-mill feeder, which is a combination direct and scoop feeder. The main ore stream passes through the center of the feeder directly into the Hardinge 7-foot by 36-inch ball mill. The return sand product from the drag classifier, together with the middlings of the flotation cleaner cell and other material sent back for regrinding, go to the feeder box, whence they are scooped up by the feeder at each revolution of the ball mill.

Formerly all feed to the ball mill passed to the feeder box, and all had to be scooped up by the feeder scoop. Considerable trouble was experienced while this method of feeding was used, due to the coarseness of the feed and the fact that all of the feed had to pass through the scoop, which in turn caused excessive wear on the scoop and the shearing of the bolts which held the feeder to the ball mill.

The ball mill is driven by a 100-horsepower motor through a special link-chain drive and jack shaft which transmits the power by a beveled gear to the master drive of the ball mill. A special switch and rheostat are used in starting the motor, so that the load may be taken up gradually. The mill operates at a speed of 26 revolutions per minute, and it is of utmost importance that this speed be maintained. If for any reason the speed of the mill is reduced, coarse ore starts to accumulate in the mill, throwing a greater load on the motor, which in turn retards the speed of the mill. It has been found necessary to reduce the rate of ore feed to the mill when the speed is reduced 1 revolution per minute and to stop the mill when the speed is reduced to 20 revolutions per minute.

The density of the pulp in the ball mill is maintained between 1.76 and 1.88. Tests made at this concentrator indicate that if enough water is used to obtain a free-flowing pulp the less water used in the ball mill the less the wear on the liners, the less the ball consumption, the larger the capacity, and the more uniform the ground product. Another important advantage in a thick ball-mill pulp is the ease of pulp-density control in the classifier, since most of the water is added at the discharge end of the ball mill, and any change in moisture content in the ball mill can be quickly taken care of by changing the amount added to the pulp as it flows into the classifier. The importance of this advantage is more evident when starting the mill and at times when floors are washed and the pump drained into the ball-mill feed box. Density control has proved to be of so great importance that an indicator is kept constantly in the pulp at the overflow end of the ball mill, and this indicator is in plain view of the operator from almost any position in the concentrator.

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During the first part of the period covered by this paper special chrome-vanadium forged-steel balls ranging from $2\frac{1}{2}$ to 5 inches in diameter were used. An average charge of balls added at one time consisted of 200 pounds of $2\frac{1}{2}$-inch, 200 pounds of 3-inch, and 200 pounds of mixed 4-inch and 5-inch sizes. The ball load amounted to 27,000 pounds. These balls gave good service, and during the period they were used the ball consumption amounted to 2 pounds per ton of ore ground. After the intermediate crushing unit reduced the ball-mill feed to $\frac{1}{2}$-inch size the larger balls were found too large for best results, in that they caused more liner replacements, and with the finer size of feed they have the tendency to wear irregularly, which causes them to slide rather than roll as they descend from the upper side of the ball mill. At present cast-iron balls are used in sizes ranging from 2 to $3\frac{1}{2}$ inches. The consumption of balls in pounds per ton of ore ground has increased slightly, since the use of forged-steel balls has been discontinued, but considered from the cost standpoint the cast-iron balls are more economical.

Cast-iron liners are used, and since one set of liners lasted the entire period covered by this paper definite figures on liner consumption can not be given. Judging by the condition of the liners at the end of the mill, however, it was apparent that replacements would soon be necessary, and upon this basis the liner consumption would amount to approximately $\frac{3}{4}$ pound per ton of ore ground.

The capacity of the ball mill, amounting to 200 tons of ore ground per 24 hours, is higher than that obtained at other concentrators where similar sizes of ore feed and ground product prevail. The high capacity is due to the absence of talc or other substance in the ore that has a cushioning effect within the ball mill.

The ball mill is in closed circuit with a $4\frac{1}{2}$ by 16 foot Dorr drag classifier set on a slope of 22°, the rakes operating at a speed of 20 strokes per minute. When the ball mill and classifier circuit is operating at full capacity and when the density of the classifier overflow is maintained at 1.56 the tonnages and sources of materials entering the ball mill are as follows:

Dry ore	tons per hour	9.0
Return sands from classifier.	do.	3.2
Flotation middling from cleaner cell	do.	2.0
Total solids	do.	14.2
Water entering ball mill	gallons per hour	1,762
Solids in ball mill pulp, calculated	per cent	66

Table 1 gives the screen sizes of the concentrator head and the various intermediate and final products of crushing and grinding for the period February 15, 1929 to June 1, 1929.

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Table 1. - Screen sizes of crushed and ground products

Screen size	Mine run-feed to coarse crusher	Coarse crusher product	Ball mill		Drag classifier	
			Primary feed from rolls	Secondary feed, classifier sands and flotation middling	Feed	Overflow
Plus 3-inch ring	50					
Minus 3-inch plus 2-inch ring	15					
Minus 3-inch ring plus 1-inch square	13	30				
Minus 1-inch square plus 4 mesh	8	25	35			
Minus 4 mesh plus 10 mesh	5	20	30			
" 10 " " 70 "	4	15	20	35	10	
" 70 " " 200 "	5	10	15	60	39	35
" 200 "	-	-	-	5	51	65

Flotation

The classifier overflow is divided between two No. 12 K & K rougher flotation machines. The concentrates from the head end of the rougher cells, amounting to approximately 15 per cent of the total rougher concentrates, are a finished product and goes directly to the filter. The remainder of the concentrates from the two roughing machines goes to one No. 12 K & K cleaner machine. The cleaner flotation machine produces finished concentrates, which go directly to the filter, and a middlings product, which is returned to the ball mill for additional grinding.

The K & K flotation machine produces a froth by means of a mechanical agitator or rotor. The No. 12 machine agitator chamber is 2 feet in diameter and 12 feet long. The rotor which revolves in this chamber is 20 inches in diameter and 11 feet 8 inches in length. The pulp enters the machine near the lower corner of the agitation chamber nearest the frothing compartment, and air is drawn in around the rotor shaft at both ends of the cell. A standpipe at the intake allows for a head of nearly 2 feet of pulp.

The action of the cell is as follows. Pulp enters the intake, is picked up by the action of the rotor, and is forced to the circumference of the rotor where it is mixed with air drawn into the cell by suction created by the displacement of the pulp. The pulp travels back along the bottom of the cell and forward over the top, and at the front of the chamber it strikes a baffle which cuts part of the pulp stream out and guides it through a perforated wooden

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partition to the frothing compartment. Gates at the bottom of the frothing chamber allow for the adjusting of the flow of pulp from this compartment back into the agitation chamber.

A particle of ore therefore travels a spiral route through the cell, coming into the frothing cell again and again. Practically all material that can be floated can be taken from the pulp in the rougher cell, due to the length of the cell and the close adjustments possible.

The pulp traveling downward from the perforated partition to the gate in the frothing chamber has a scrubbing effect upon the upward moving concentrates. Because of this action the minerals tend to separate; the finest chalcopyrite floats first, coarser chalcopyrite and fine pyrite next, and coarser pyrite last. If the ore has not been ground finely enough some copper minerals will float with the pyrite, due to the dampening effect of the particles of gangue which adhere to the chalcopyrite.

Care is exercised by the operator to keep the level of the pulp at such height that all copper minerals float and go over the edge of the frothing chamber in the first 9 feet of the cell. The tailings end of the chamber is free from copper minerals for a distance of 3 feet. This part of the frothing chamber acts as a safety on cell operation. Any small discrepancy in grade of ore, density of pulp, and reagent feed can be taken care of for a short time without loss of copper or lowering the grade of concentrates.

The cleaner cell is so adjusted that high-grade concentrates are obtained throughout the length of the cell.

Carefully conducted screen tests on the tailings product of the cleaner cell proved that it is coarser than the concentrates although the difference is not great. Microscopic observations showed that many of the particles in this product were made up of chalcopyrite, pyrite, and gangue or some combination of two of these materials.

These tests and observations made it evident that regrinding of the cleaner cell tailings was necessary if a high recovery and clean concentrates were to be made, and this product is therefore returned to the head of the ball mill.

The high recovery of copper, amounting to 99.2 per cent, is due to a large extent to this procedure. Regrinding of the cleaner cell tailings not only allows for the recovery of the copper in this product but gives the operator more leeway in the control of the cleaner-cell operation. The operator realizes that any values that go into the cleaner tailings will be recovered subsequently; therefore the concentrates can be skimmed cleaner without danger of lowering recovery. Under these conditions high-grade concentrates are produced as well as high recovery obtained.

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As previously described, the Harmony ore contains chalcopyrite so intimately mixed with pyrite that fine grinding is necessary to free the former. There is a slight difference in the flotability of the two minerals without the use of a depressent. This difference is taken advantage of in the present milling practice, and it is considered just as important to depress the pyrite into the tailing as to cut down the silica content of the concentrates.

Test runs at the concentrator proved that if enough collector oil is added at the head of the ball mill and sufficient frothing oil is added at the feed end of the rougher cell it is possible to float practically all of both minerals. It was also determined that when insufficient collector oil was added to wet both chalcopyrite and pyrite it had a slight preference for chalcopyrite. Other phenomena that became apparent during the tests were: Excessive amounts of collector oil increased the pyrite content in the concentrates; excessive amounts of frothing oil increased both pyrite and silica content in the concentrates; insufficient amounts of either reagent caused losses of copper in the tailings; acidity or alkalinity had little effect on results; and density of pulp going to the cells must be kept constant and within very narrow limits.

The problems facing the operator are therefore to control the feed to the ball mill so that all of the ground product will pass through a 70-mesh screen, to keep the density of the feed to the rougher cells between 1.526 and 1.559, to add correct amounts of collector and frother oils, and to control the height of pulp in cells so that a clean separation is made.

The amount of reagent necessary was originally determined by tests, so it is obvious that the exact amount to be used under any set of conditions can only be approximated. The most noticeable feature of the process, however, is the small amount of reagents used.

When the concentrator treats 200 tons of 4 per cent copper ore per 24 hours the collecting oil is added at rate of 18 drops per minute to the ball mill, and the frother is added at rate of 30 drops per minute to the pulp as it enters the rougher cell. This amounts to 0.2 pound of collecting oil and 0.6 pound of frothing oil per ton of ore treated.

In the frothing compartment of the rougher cell there is a gradation in the floated product from pure chalcopyrite at the feed end to pyrite at the tailing end. It has been found practicable to obtain clean concentrates from near the head of this cell. Sheet-iron lap strips are used to convey this product over the regular launder to a special launder that runs directly to the filter.

When the rougher cell is carefully operated the cleaner cell needs little attention. The main object is to get clean concentrates. A slight variation in the amount or grade of cleaner tailings sent to the ball mill for regrinding makes little difference in mill operation.

In test runs it was noticeable that agitation of the pulp between the ball mill and the rougher cell made it almost impossible to keep the tailing end of the rougher cell entirely free from chalcopyrite. Additional reagents and the raising of the pulp level in the cell had an unsatisfactory effect on cleaner cell operation.

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As the machinery is now arranged there is no agitation at all between the ball mill and the flotation cells, and cell operation is almost perfect.

A screen with $\frac{1}{4}$-inch holes removes all debris from the pulp as it flows from the ball mill, and a short launder connects the classifier with the rougher cell.

Table 2 gives the screen sizes and Table 3 the tonnages and analyses of flotation head and flotation products for the period February 15, 1929 to June 1, 1929.

Table 2. - Screen sizes of flotation feed and products

Screen sizes	Rougher machines			Cleaner machines		
	Heads	Concentrates	Tailings	Heads	Concentrates	Tailings
Minus 70 mesh plus 100 mesh	10	4	11	4	2	5
Minus 100 mesh plus 150 mesh	10	12	11	12	6	12
Minus 150 mesh plus 200 mesh	15	12	15	12	12	12
Minus 200 mesh	65	72	63	72	80	71

Table 3. - Tonnages and analyses of flotation head and products

Product	Tons per hour	Copper Per cent	Iron Per cent	Sulphur Per cent	Silica Per cent	Alumina Per cent	Lime Per cent	Gold Oz. per ton	Silver Oz. per ton
Rougher flotation									
Heads	11.0	4.59	14.56	14.30	58.27	4.03	3.47	Tr.	0.14
Concentrates	3.2	15.70	30.00	28.70	21.20	2.35	1.80	.01	.50
Tailings	7.8	.02	8.22	9.10	73.30	4.71	4.14	Tr.	Tr.
Cleaner flotation									
Heads	3.2	15.70	30.00	28.70	21.20	2.35	1.80	.01	.50
Concentrates	1.2	28.61	33.56	32.40	4.90	.08	.04	.02	1.04
Tailings	2.0	7.96	27.86	26.50	31.00	3.70	2.90	Tr.	Tr.

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DEWATERING AND HANDLING OF CONCENTRATES

The concentrates from the heads end of the rougher cell and all concentrator from the cleaner flotation cell flow directly into an Oliver continuous filter having a drum 8 feet in diameter by 4 feet wide. The filter, of 50 tons capacity in 24 hours, operates 16 hours per day, and when idle the concentrates in it are agitated by mechanical agitation or by forcing air into the pulp at the bottom of the filter tank. The concentrates are scraped from the filter surface and fall on a belt conveyor which carries them to the concentrates storage bin.

A thickener ahead of the filter would undoubtedly improve filter performance, but since the capacity of the filter is ample without the thickener there seemed no necessity for installing it.

The dry vacuum pump is designated as a 12 by 14 inch compressor and is operated by a 30-horsepower motor.

Previous to the renovation of the mill Aikins' classifiers were used to dewater the concentrates. The dewatered product containing 9 to 11 per cent of moisture was sloppy, causing difficulty in handling. The overflow of the classifiers, which was pumped back to the flotation cells, contained the finest portion of the concentrates which had an undesirable effect on cell operation. It was found difficult to maintain the correct pulp density in the cells and the variation in flotation reagents caused trouble.

When the filter was first installed a centrifugal pump was used to remove the water from the vacuum receiving tank. The pump discharged the water into the main water-supply tank of the mill. This arrangement was satisfactory unless the filter cloth leaked and let concentrates into the filtrate. The wear on the pump and pipe line was also excessive.

A settling tank was installed at an elevation of 35 feet below the filter and a pipe run into it from the bottom of the receiving tank. The lower end of the pipe line is about 1 foot below the surface of the water in the settling tank when the tank is overflowing. The water becomes a seal when the force induced by the vacuum causes water to rise in the pipe. Filter performance improved materially with this arrangement. Water which overflows is always clear, and during seasons of low water this overflow is pumped into the supply tank of the mill while at other times it is allowed to run to waste.

Small leaks in the filter cloth do not give trouble, since the water does not pass through a pump, nor do leaks cause loss of concentrates since concentrates that get into the filtrate are recovered in the settling tank.

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The following tabulation gives data on filter operation.

Density of concentrates fed to filter		1.30
Moisture in feed to filter	per cent	51
Moisture in filter cake	do.	6
Total filter surface	square feet	100
Filter capacity	tons per 24 hours	50
Filter capacity	tons per square foot per 24 hours	0.5
Time operated	per cent	66
Life of filter cloth	days	110
Tons treated per square foot of filter cloth		33
Vacuum maintained	pounds per square inch	6.2
Air pressure used on blow	pounds	5.0

DEWATERING AND DISPOSAL OF TAILINGS

Tailings from the rougher flotation cell are run by launders to two cone-type classifiers at the lower end of the tailings pond. A relatively thick product containing approximately 18 per cent of moisture drops through the spigot of the classifier and gradually builds a dam across the tailings dump. To distribute the thickened tailings the classifiers are mounted on a truck that runs along a broad track. The track is supported by scaffolding which has its foundation on the floor of the canyon, the scaffolding being gradually buried and new units added above. The classifiers remain at one level until the dam is built as high as the spigot outlet. This is done by moving the classifiers a few feet after the thickened pile has been built up to the spigot. As the dam gets higher the pond holds more and more tailings until now one raising of the classifiers will raise the dam to a height sufficient to impound more than 100,000 tons of tailings.

The overflow from the classifiers flows by launders to a point 20 feet back of the dam and falls into the pond. In the quiet water of the pond the sediment even to the finest slime soon settles. The clear water is carried under the dam by a trough, the intake of which is at the back of the pond and the discharge on the canyon floor at the foot of the dam.

During a few months in winter a portion of the tailings is filtered and the water reclaimed. A Portland continuous 8-foot by 8-foot filter is used for this purpose. This filter is operated by the same vacuum system as the concentrates filter. Preliminary preparation is not necessary when the filter is to be operated, because it is always connected with the receiving tank of the vacuum system. The operator has simply to send a portion of the tailings to the filter and open the vacuum check valve, and the filter operates without further attention.

The vacuum system has sufficient capacity to take care of both filters.

A belt conveyor 60 feet long carries the filtered tailings from the filter to the tailings dump.

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It is not important for the tailings to be low in moisture, therefore the vacuum is cut off from nearly half of the drum above the filter tank, thus giving a larger capacity and maintaining a higher vacuum pressure.

SAMPLING AND OPERATING DETAILS

The heads sample is taken at the overflow of the classifier, as the pulp drops into the launder which carries it to the flotation cells. A cutter moves across the pulp stream at intervals of 15 minutes, taking about one second to cross the stream. A sample weighing 10 pounds is obtained during each 8-hour shift. This sample is dried on a steam plate and further cut down by a riffle to a sample weighing one-half pound.

Concentrates are sampled at 30-minute intervals at the filter by cutting a groove one-half inch wide through the filter cake on the drum. The weight of concentrate sample amounts to 5 pounds for each 8-hour shift, and this sample is cut to one-half pound before being taken to the assay office.

The concentrates sample is taken as an aid to mill operation and is not considered as control data on shipments. All shipments of concentrates are sampled at a custom sampler.

Tailings are sampled by an automatically operated cutter sampler, the cutter, which is 1 inch wide, moving across the tailings stream at intervals of 15 minutes. A tailings sample weighing approximately 25 pounds is secured for each 8-hour shift and after drying this sample is cut down by a riffle to a sample weighing one-half pound. The mill operator takes all samples and places them on the steam plate, and the assayer cuts them down to the assay size.

To insure against any changes that may come about in any of the operating units or in the ore or the reagents the operator is required to inspect every part of the grinding and concentrating units every half hour. On these tours of inspection he makes the following observations and records them on a special form which is taken to the office at the end of each shift:

Height of gate on automatic ball mill feed.
Speed of ball-mill feed belt.
Amount of collector reagent added in drops per minute.
Amount of water flowing into ball-mill feed box.
Speed of ball mill.
Density of classifier overflow pulp.
Speed of flotation-cell rotors.
Appearance of froth on all flotation cells.
Amount of frothing reagent added.
Time between cuts of the mechanical samplers.
Thickness of filter cake.
Amount of vacuum at vacuum receiving tank.
Approximate amount of tailing flowing from the cleaner cell to sump.

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Oiling of bearings and machinery is done by the compressor man, who handles this work in addition to looking after the compressor, which is in the mill building and which supplies the mine with air.

The superintendent obtains the assays of the heads for the night shift, and from these assays he estimates the amounts of reagents that are to be added for that grade of ore. As long as a noticeable change does not take place in the ore coming to the mill these estimates are considered nearly enough correct to be used during the day. The reagents feeders are therefore set by the superintendent each morning and merely checked over during the rest of the day.

When it is obvious that the grade of ore has changed materially a hand sample is taken from the fine-ore storage bin. This sample is assayed at once to determine the changes necessary in the amounts of reagents required. Until the result of this assay is known adjustments are made at the flotation cells to take care of the variation in the grade of the ore. When the assay results are known adjustments in amounts of reagents are made immediately, and within a few minutes the mill is operating under the new conditions. The general average of ore sent to the mill is kept as nearly uniform as is possible, and readjustment of the feeders during the day is rare. Assays of concentrates and tailings also help in determining the correct amount of each reagent that is required.

WATER SUPPLY

Water is obtained from a spring 500 feet distant up the canyon from the concentrator and flows to the concentrator through a 2-inch pipe line. The supply is adequate for 8 months in the year at capacity operation, but during the other four months water in varying amounts must be reclaimed. The amount reclaimed varies from a few hundred gallons per day to 9,000 gallons per day, the latter figure amounting to nearly one-third of the total amount used in the mill. This larger amount is reclaimed only during periods of constant cold weather, when the spring gets very low and water is not running on top of the ground.

The mill operating at full capacity requires about 33,000 gallons of water per 24 hours. This amount gives a pulp density of 1.56 and allows for some loss. The water is exceptionally pure and needs no conditioning whatever.

CONCENTRATOR RECOVERY AND METALLURGICAL DATA

The recovery of copper for the period February 15, 1929, to June 1, 1929, amounted to 99.2 per cent in concentrates averaging 28.61 per cent of copper.

Table 4 gives metallurgical data for period February 15, 1929 to June 1, 1929.

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Table 4. - Metallurgical data

Total ore treated	tons	17,000
Total operating time	days	85
Operating time per day	hours	24
Ore concentrated per 24 hours	tons	200
Sections operating		1
Concentrates produced	tons	2,260
Average weight of concentrates produced in 24 hours	do.	28.2
Recovery of copper	per cent	99.2
Oxide copper in ore	do.	None
Ratio of concentration ... tons of feed to tons of concentrates		7.5
Net water consumption per ton of ore treated	gallons	106
Ball consumption per ton of ore treated	pounds	3.1
Liner consumption per ton of ore treated	do.	0.5

Table 5 summarizes concentrator costs for the period February 15, 1929 to June 1, 1929. Table 6 summarizes concentrator costs in units of labor and power during the same period. During this period mill operators received $5.00 per shift and other labor received 50 cents per hour. Power cost is 1.4 cents per kw. hr.

Table 5. - Summary of concentrator costs, dollars per ton of ore concentrated

Operation	Operating labor	Power	Supplies other than reagents	Reagents	Repairs		Miscellaneous	Total
					Labor	Supplies		
Breaking to 1½ inch size	$0.015	$0.007	$0.005		$0.005	$0.005		$0.037
Intermediate crushing to ½-inch size	.015	.007	.005		.005	.005		.037
Grinding to 65 per cent minus 200 mesh size	.040	.070	.140		.020	.014		.284
Classifying, screening, and conveying	.020	.008	.012		.002	.010		.052
Flotation concentration	.030	.013	.010	0.017	.010	.005		.085
Dewatering and handling concentrate	.015	.010	.010		.012	.010		.057
Sampling and assaying	.015		.002					.017
Tailings disposal	.010							.010
Pumping and magnetic pulley	.005	.002	.005		.003	.008		.023
Miscellaneous	.010	.003	.005		.011	.010	.010	.049
Total	.175	.120	.194	.017	.068	.067	.010	.651

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Table 6. - Summary of concentrator costs in units of labor and power

	Labor, man-shifts of 8 hours, per ton of ore concentrated			Power	
	Operation	Repairs	Total	Kw. hr. per ton of ore concentrated	Per cent of total
Breaking	0.0030	0.0010	0.0040	0.70	4.4
Intermediate crushing	.0030	.0010	.0040	.70	4.4
Grinding	.0080	.0040	.0120	8.00	50.0
Classifying, screening and conveying	.0040	.0004	.0044	.80	5.0
Flotation concentration	.0060	.0020	.0080	2.30	14.4
Dewatering and handling concentrate	.0030	.0024	.0054	3.00	18.8
Tailing disposal	.0020	- -	.0020	- -	
Water supply including pumping	.0010	.0006	.0016	.20	1.2
Assaying and sampling	.0030		.0030	- -	
Miscellaneous	.0020	.0022	.0042	.30	1.9
Total	0.0350	0.0136	0.0486	16.00	100.0

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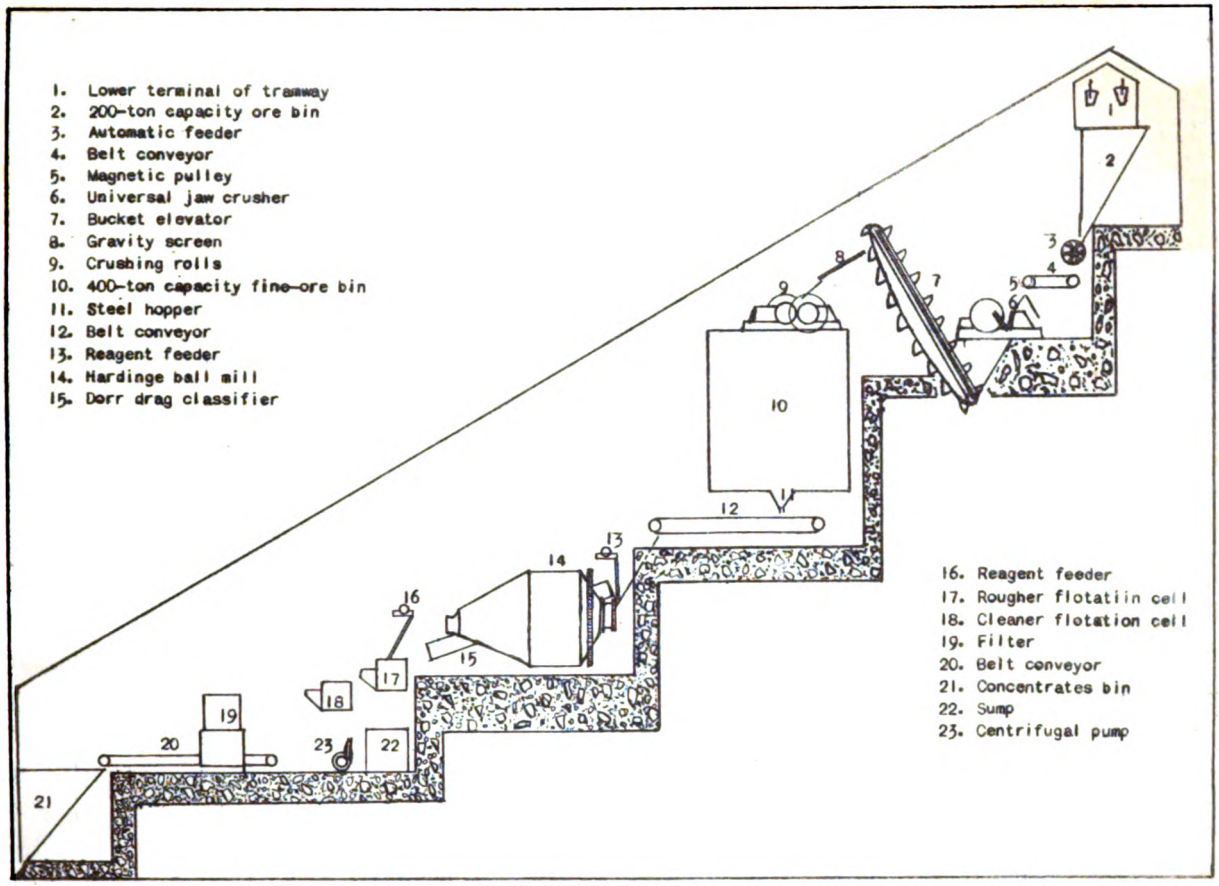

Figure 2.- Side view of concentrator showing position of machines

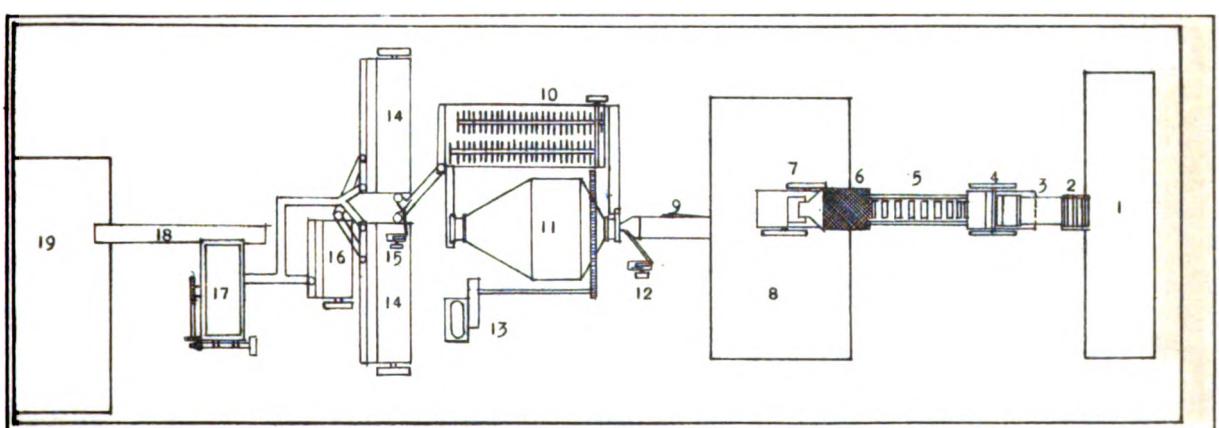

Figure 3.- Plan of concentrator

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